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Modeling Energy Consumption in the Mining and Milling of Uranium

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Modeling Energy Consumption in the Mining and Milling of Uranium

by

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to dreaming big and always looking forward

one love

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Modeling Energy Consumption in the Mining and Milling of Uranium

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A family of top-down statistical models describing energy consumption in the mining, milling, and refining of uranium are formulated. The purpose of the models is to estimate the energy-to-grade dependence for uranium extraction, while defining a minimum grade that can be feasibly mined and produced. The results serve as a basis for understanding the factors governing energy consumption in the production of U_3O_8. The models are applied to a considerably larger data set of operating mines than in any previous effort. In addition, the validity of the modeling approach is established by modeling energy for two other commodities, gold and copper, thereby showing it can be applied to other metals.

Statistical measures of explanatory power show that the models the energy-to-grade relationship is well-described for both uranium and gold. For copper, there was insufficient data over a broad range of ore grades to obtain a model that passed statistical confidence measures. The results show that mining of lower-grade deposits of uranium is likely to be less energy-intensive than previous investigators concluded. It is shown that the uncertainty in the results is dominated by the contribution of the grade-independent component of energy consumption.

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Chapter 1

Introduction

As the nuclear industry enters a new era of growth, two hotly discussed topics are life cycle energy analysis and carbon emissions. These issues will help shape future choices for consistent and reliable base-load power sources. However, as a recent review [Sovacool 2008] of carbon emissions associated with the nuclear industry illustrated, great disagreement has been found between carbon emission estimates for nuclear fuel cycles. It is noteworthy that emissions are clearly strongly influenced by the front-end processes: the largest disagreement is found in front-end portion of Table 1.1, which includes mining, milling, conversion, enrichment, and fuel fabrication. The most energy intensive processes in the front-end are mining, milling, and enrichment. There is no doubt that the values for the frontend in Table 1.1 are based on the energy use in these steps. The vast difference in values for the front-end brought into question the methods behind the results reviewed in [Sovacool 2008] and led to this study.

Table 1.1. Summary of CO_2 emissions arising from the nuclear fuel cycle, source: [Sovacool 2008]

[kg CO_2/ MWh(e)]	Frontend	Construction	Operation	Backend	Decommissioning	Total
Min	0.68	0.27	0.1	0.4	0.01	1.36
Max	118	35	40	40.75	54.5	288.25
Mean	25.1	8.20	11.6	9.2	12.0	66.1
% of total	38%	12%	18%	14%	18%	100%
N	17	19	9	15	13	—

Consultation of the literature associated with Sovacool and others showed that values for the energy consumption in mining, milling, and refining were often not supported by consistent assumptions or primary data from operating mines. Chapter two covers the literature review in detail.

While reviewing papers, it was seen that energy consumption from mining and milling was usually modeled as having an inversely proportional dependence on the grade of the uranium ore that is mined, but that the proportionality constants differed greatly. Grade is defined as the mass of commodity contained in a certain amount of ore. Therefore, a main objective of this report is to quantitatively study the energy-grade relationship for mining uranium and producing U_3O_8.. These processes become more energy intensive at lower grade ores because more waste rock must be removed to obtain the uranium ore. Likewise in milling, more energy is needed to separate the uranium ore from the mined orebody because there is less uranium in a given rock formation. In order to model these processes, data would have to be collected. It was essential to gather as a full data set as possible from actual mining companies that included energy consumption, mass of waste rock removed, mass of ore milled, and mass of product. These data are fitted to linear least squares regression models to define energy intensity coefficients for each of the three throughputs mentioned.

This approach to understanding fundamental dependencies of energy use in mining was first taken by [Chapman 1975], but the current study collects and applies a greatly expanded data set. To confirm the validity of the regression models applied to uranium, this report also collects mine data for other metal commodities and subjects them to the same statistical analysis. Although other metals use different mining and milling techniques, the energy to ore grade relationship should hold for all mined commodities. In this report gold and copper, both with similar mining and milling techniques to uranium, were selected for this comparative study.

The following report should engage those persons interested in fuel cycle energy consumption. This report will first outline the review of useful literature that was studied prior to the production of this document. This will lay the foundation for chapter three, where the methods and approaches will be described in detail. The results obtained when these methods are applied to uranium, gold and copper will be shown and discussed in chapter four.

Conclusions will be drawn at the end of this report, and an appendix with all data used for the analysis can be found thereafter.

Chapter 2

Literature review

This chapter highlights the progress that has been made in the nuclear industry to understand energy consumption in the mining, milling, and refining of uranium. This chapter begins with an overview of historical ore grade trends and is then followed by the work done by previous investigators. The review of related work serves as a background and foundation on which this report is based and will be outlined in this chapter.

2.1.

In the nuclear energy industry the question of sustainability is being raised in connection with the long-term supply of uranium. Assertions have been made as to how long the uranium supply will last, based on current reserves. A common theme in the discussion is the question of when will the mining of uranium become endothermic, that is when will more energy be expended in recovering uranium ore than energy that goes to the grid from the nuclear reactor. This question entails a look at the past, current, and future ore grade of uranium.

Historically, the highest-grade deposits have been exhausted first because these require the least amount of energy to mine, are easiest to find, correspondingly are the lowest costs to companies. For example, a commodity with global importance is gold and its historical world-averages for ore grade are given in Figure 2.1. The world averages are from countries for which the data was available: the US, Australia, Canada, Brazil, and South Africa. This trend line clearly shows that ore-grade has been decreasing over time, and therefore motivates this report's ultimate goal of understanding the energy-to-grade relationship. The peaks in Figure 2.1 describe gold rushes pertaining to different countries.

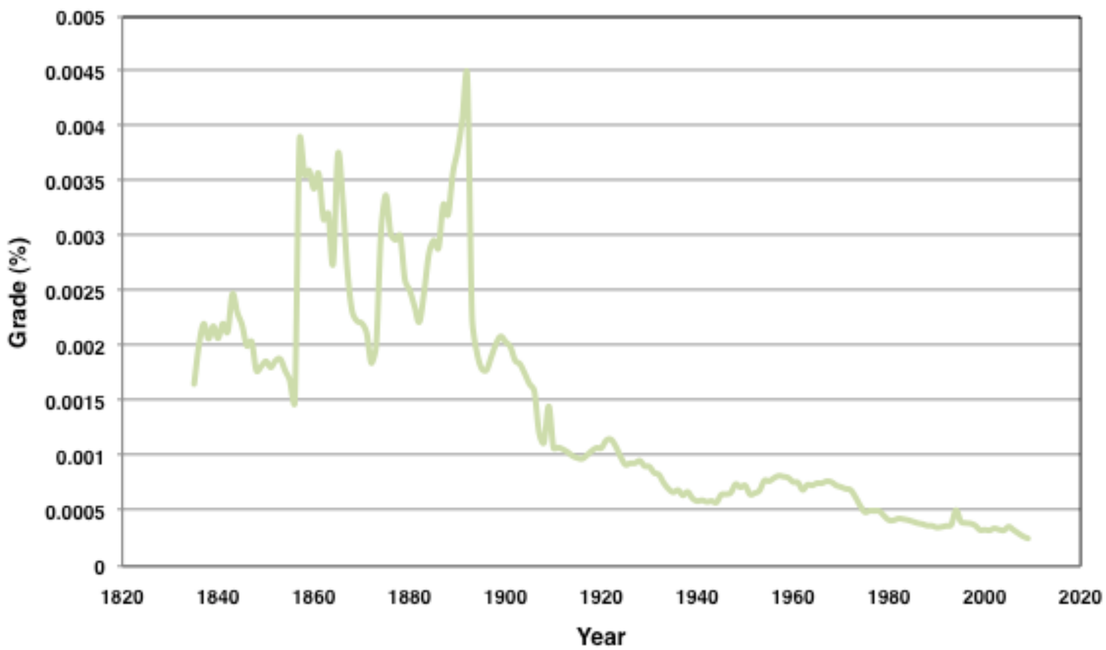

Figure 2.1. History of world-averages for gold grade (%) [Mudd 2010a]

Likewise, Figure 2.2 shows the trends for copper, uranium, and nickel in percent grade versus time. The same key trends are noted for these three other commodities. The following figure was taken from [Mudd 2009] and the peaks and valleys are noted on the top of the figure. It should be noted that ore grade is given in percent for copper and nickel, but as kg/t for uranium.

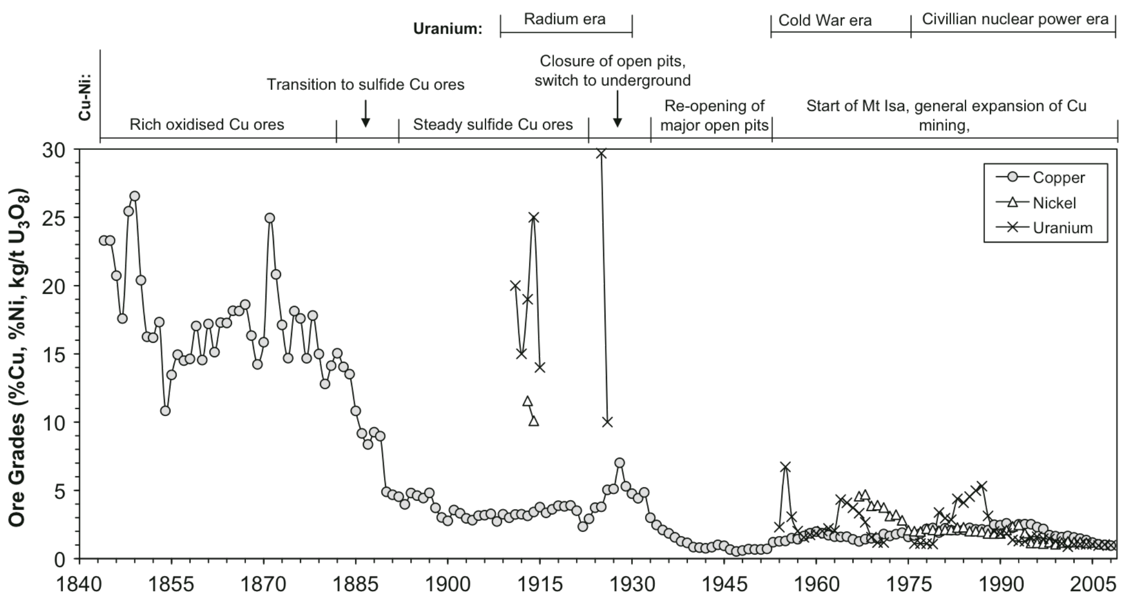

Figure 2.2. History of world-averages for copper, uranium, and nickel grades, Source: [Mudd 2009]

2.2.

The first attempt to create a generalized model of energy use in uranium mining was by Peter Chapman. Chapman's primary study was done "to investigate the effect of the grade of uranium ore on the viability of thermal reactor systems. . .by an analysis of the energy to produce copper which showed that the energy required was inversely proportional to the grade of ore" [Chapman 1975]. Chapman was the first to notice the trend that decreasing ore grade has on energy consumption and exemplified this with theoretical energy numbers from copper mining, milling, and refining [Chapman 1974]. Chapman's formulation of the energy required to produce a tonne of refined product incorporated contributions from the mining, milling and product refining steps.

Since Earth's known resources are being slowly exhausted, Chapman wanted to answer at what point mining uranium would take more energy than the uranium ore would produce in a nuclear reactor. Chapman used real and hypothetical data to model energy consumption at the mine and mill. By including a 1/G term for the grade of the ore, Chapman depicts the relationship between grade and energy use as being inversely proportional.

In order to quantitatively describe the reviewed models, it is necessary to identify all terms that will be used (Table 2.1).

Table 2.1. Quantities used in reviewed models

Symbol	Unit	Description
e	(GJ(e) +GJ(t)) / tU	Energy required to produce 1 tonne of refined U (as U_3O_8)
e_{mine}	(GJ(e) +GJ(t)) / (tonne of ore + overburden)	Energy required to mine one tonne of material
e_{mill}	(GJ(e) +GJ(t)) / (tonne of ore)	Energy required to mill one tonne of ore
e_{refine}	(GJ(e) +GJ(t)) / tU	[Chapman 1975] interpretation: "energy required to convert beneficiated ore to required material"
$e_{product}$	(GJ(e) +GJ(t)) / tU	[Prasser 2008] and current document interpretation: As [Chapman 1975] above, plus other energy inputs not directly proportional to the masses of mined material or ore
G	% U_3O_8	Ore grade
S	kg overburden/kg ore	Stripping ratio
Y	kg U in mill output / kg U in mill input	Ore milling yield

Chapman's 1975 model was the following:

$$e = \frac{100}{0.848G}\left(e_{mill} + (1+S)e_{\min e}\right) + e_{refine} \qquad (2.1)$$

Each of the coefficients in equation 2.1 is associated with a different mass flow that can be viewed in Figure 2.3. For example, e_{mill} , is the mass flow proportional to the amount of ore. Similarly, the quantity $(1+S)e_{mine}/G$ is the energy required to extract the necessary ore plus overburden in order to extract one tonne of mill-able uranium from the mine. The factor 0.848 converts a tonne U_3O_8 to a tonne U. Chapman also included a term e_{refine}, representing the energy needed to produce and purify yellowcake from milling product containing 1 tonne of uranium, but he did not estimate its value nor did he include it in his final formulation of the energy intensity of uranium production.

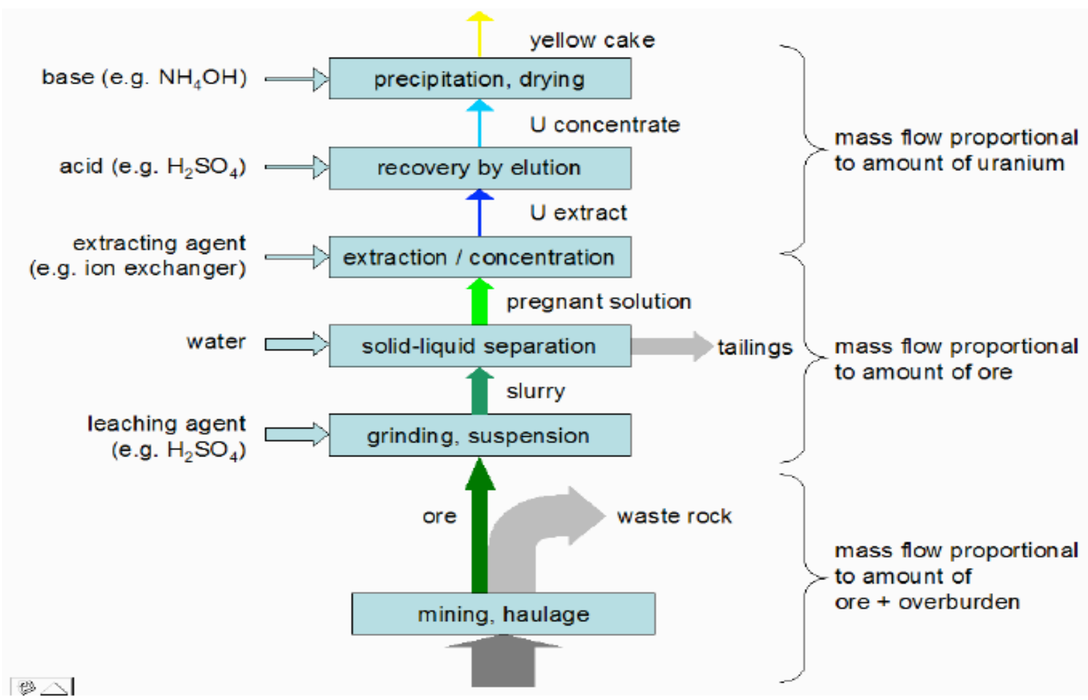

Figure 2.3. Mass flows in uranium mining and milling, Source: [Prasser 2008]

Due to the limiting factor of mine data availability, a problem still seen today, Chapman could only assemble two data points in order to define his energy coefficients. The data that he used from [Everett 1963] included four mines operating in Wyoming in the early 1960s with an average ore grade of .31% and stripping ratio of 24. Chapman weighted the mines proportional to their estimated reserves to acquire 1210 MJ(t) per tonne of ore for mining and 99 MJ(e) + 828 MJ(t) per tonne of ore for milling.

Often energy data is separated into thermal and electric carriers. At open pit mines, it is common for diesel fuel (thermal units) to run the equipment on site, whereas at the milling and refining facilities most of the energy comes from electricity (electric units). Chapman aggregated the thermal and electrical energy consumed when deriving his model coefficients. Reports to follow, as well as the body of this report, follow this method. At the mine, energy carriers may change over time (as technology for electric driven shovels etc advances), and thermal-to electric conversion efficiencies are difficult to identify, so common practice is to use aggregated energy when expressing consumption.

The second data point that Chapman used was for mining and milling of Chattanooga shale at .007% ore grade using underground mines [Bieniewski 1971]. The stripping ratio is assumed zero because for this hypothetical situation the ore body was considered the entire shale formation (no overburden).

Although energy use for both data sets comes primarily from direct energy use from fuel and electricity, embodied energies, or energy from the consumption of chemicals, machinery, and machine parts, were also considered in Chapman's analysis [Chapman, 1975]. In practice embodied energy only constitutes about 10-20% of energy, or 1-4% of total energy costs [Chapman 1974]. Analysis similar to what Chapman did for the US mines was again done for the Chattanooga shale mine to acquire energy inputs for mining and milling the ore compiled in Table 2.2.

Table 2.2. Chattanooga shale data

	GJ(e)/t U	**GJ(t)/t U**
Mining, S=0	32.7	36
Milling	77.5	21.95

At this point is unclear what Chapman did in his analysis. After aggregating the energy inputs for mining and milling from Chapman's paper, 0.81 TJ(t+e)/tU for the US mines and 30.2 TJ(t+e)/tU for the shale project, are found. However, Chapman fitted his model using 1.0 and 20.0 TJ(t+e)/t U for the 2 mines. His reasoning is vague in the 1975 paper, but his coefficients based on equation 2.1 are summarized in Table 2.3. Using his model, the electrical energy yield from mining a tonne of uranium is 108 TJ(e)/tU. Using this, Chapman arrived at the cut-off grade, G=.002%., or the ore grade uranium would have to be in order for the energy input for mining and milling to equal the energy output from the uranium product.

Table 2.3. Coefficients of Chapman model

e_{mine}	0.071 GJ(t+e)/t ore + overburden
e_{mill}	1.329 GJ(t+e)/t ore

Chapman's model led to other attempts by following investigators to predict the ore grade at which the process would become endothermic. It is noteworthy that Chapman ended his 1975 paper with, "Clearly a lot more data on different uranium mines, with different ore grades, different rock hardness etc is needed to substantiate this estimate."

There were two modifications to Chapman's model for metal extraction reported in [Rosa 2008] by Kellogg in 1977. Kellogg, as referred to in [Rosa 2008] first stated that energy consumption depends on mining method and that milling will depend on ore hardness as well as leaching extraction agent at the mill. He then incorporated a yield function, Y, into the model. Ranging between 0 and 1, Y describes the recovery efficiency of the product in the milling and refining processes. If the model is dependent on the type of mining method, e_{mine} and e_{mill} coefficients will vary by method. In practice, it is difficult to implement when operating data is limited. It can also be difficult to use in a predictive capacity, as it requires additional forecasting of the mining strategies, ores to be milled, and processes to be used to mill them. Nonetheless, data found for this study indicates that differences between mining methods cannot be neglected.

Chapman continues to consider the relationship between ore grade and metal extraction in [Chapman 1983] as he emphasizes the direct variation of energy use with ore grade. He stresses importance of this relationship because the history of metal mining has been one of declining ore grade. He introduces a model that not only predicts this declining ore grade, but also incorporates technological efficiency. He defines the quantity of fuel used as, ‘the energy requirement divided by the efficiency with which the fuels are used to provide the energy”. He introduces the following model, which he originally made for copper extraction,

$$e = \frac{e_o}{g\eta_1} + \frac{\Delta G}{g\eta_2} \tag{2.2}$$

Here e_o/g [GJ(t+e)/tU] is the theoretical minimum energy required for mining and milling and ΔG (GJ(t)/tU) is the change in the Gibbs free energy from converting the mill concentrate to the final product. The terms η_1 and η_2 are the efficiencies of respective processes. For metal milling and refining recovery, efficiencies of .85 and .90 are used by [Chapman 1983] as industrial averages, while assuming stripping ratios of 2.0 for open pit and 0.1 for underground.

For the first time in the literature, the refining step of the process is being accounted for. This approach is not appropriate for use as top-down approach, that which fits actual data to the model. Instead, e_o is obtained from bottom-up simulations of model mines. No recent bottom-up simulations of uranium mines were found in the literature.

Chapman claims that although production technique and technology will vary by metal, technology is similar enough to be considered equal. Chapman states, “the energy of mining depends upon a large number of factors: mining method, rock hardness, equipment used, scale of operation, distance from mine to mill, and so on. However many studies have shown there are typical values for energy, one for open pit and one for underground.” Using theoretical minimum values from [Kellogg] and [Batelle] for the three energies, along with,

$$E = \frac{1}{R_c R_s G}\left(E_c + \left(1+S\right)E_m\right) + \frac{1}{R_s}E_s \tag{2.3}$$

Chapman surmised,

For open pit metal mining

E= 400 MJ/tonne produced

For underground metal mining

E= 1000 MJ/tonne produced

Chapman defines R as the quantity of metal in the output stream divided by the quantity of metal in the input stream. The quantities E_m ,E_c ,and E_s are the energies for mining(MJ/tonne rock), milling(MJ/tonne ore), and refining(MJ/tonne produced), respectively. It should be noted that the aggregated energy here assumes 11.25 MJ(t) is equal to 1 kWh(e), a 32% efficiency. Due to fuel use for mine ventilation, water removal, haulage, and explosives, underground mining is about an order of magnitude more energy intensive than open pit mining [Mortimer 1974].

Mudd [2007c] qualitatively assesses energy balances for uranium, copper and gold, plotting ore grade vs. energy use at gold mines and finds, “for energy consumption relative to gold ore grade and annual ore throughput, inverse exponential patterns are apparent.” Mudd in several cases considers technological advancements to improve energy efficiency at the mine and mill. Mudd [2007c] concludes, without basis for how he arrived at these numbers, that the energy input for gold rangers from 30-275 GJ/ kg Au product (0.85-7.8 in GJ/oz) or from 0.02 -1.6 GJ/t ore processed. It is assumed that Mudd arrived at these values by taking the range of values from the data he collected. Mudd states the importance in noticing this does not account for embodied energy, “the energy involved in mine and mill construction, machinery, chemicals, water supply and the energy required for rehabilitation and ongoing monitoring and maintenance.” It is typical currently for only direct (operational) energy consumption to be reported by mining companies. Mudd’s assessment will be compared with this report in Chapter 4.

[Storm van Leeuwen 2005] used Chapman’s 1970s data to again look into metal extraction, particularly uranium extraction. They modified Equation 2.1 slightly, by dropping the dependence on stripping ratio. It should be noted that although they drop the S-term, they still quote Chapman in their paper, 'The varying amount of overburden to be removed per unit ore can make differences in the specific energy requirements of a factor of five with the same ore type." Thus an average stripping ratio is indirectly assumed to be embedded within e_{mine}, although this isn't specified in the paper. Leeuwen and Smith's model is

$$e = \frac{100}{0.848G}\frac{e_{\min e} + e_{mill}(o)}{GY} \tag{2.4}$$

Also differing from Chapman's model, Leeuwen and Smith show that uranium milling is less efficient at lower ore grades (G =% U_3O_8)[1]. Therefore, they adopted a mill yield factor, Y, [kg U in mill product / kg U in ore] (equation 2.5).

$$\begin{aligned} &Y = 0.98 - 0.0723 \bullet (\log_{10}(G))^2 & & G \geq 1 \\ &Y = 0.098 & if \quad & G \leq 1 \end{aligned} \tag{2.5}$$

Rather than specifying mining strategies, Leeuwen and Smith include a dependence on milling energy with ore hardness, the term $e_{mill(o)}$ changes with soft and hard ores. Leeuwen and Smith define soft ores as those that are easy

[1] A note regarding units: ore grades are typically expressed in weight percent of metal in the ore or weight parts per million. An example of this notation is the: 1.0 w/o U as U3O8 = 10,000 weight parts per million U are simply written as 1.0% U3O8 and 10,000 ppmU, respectively, and is conventional in the literature.

to mill, with typical grades ranging from more than 10% down to about 0.01% U_3O_8, and hard ores as quartz pebble conglomerates and granites, with grades varying typically from about 0.1% down to about 0.001% U_3O_8, or less.

Leeuwen and Smith's data comes from one source, a paper published by Rotty [1975] which collected data reported by US mines to the US Bureau of Mines in 1973. The coefficients that Leeuwen and Smith found are in Table 2.4. Their estimate includes embodied energy.

Table 2.4. Model coefficients for equation 2.4

Coefficient	Value & Units
e_{mine}	1.06 GJ(t+e)/tonne ore
e_{mill}(soft)	1.27 GJ(t+e)/tonne ore
e_{mill}(hard)	4.49 GJ(t+e)/tonne ore

Leeuwen and Smith compare their estimate to that of one they made specifically for the Ranger mine, but it doesn't appear that they used actual operating data from that mine because their estimate is almost double the direct energy consumption, 355 GJ(t+e)/tonne U_3O_8, compared to 191 GJ(t+e)/tonne U_3O_8 that is actually reported at Ranger, over the same time period, 2005-2006 [Mudd 2008a],[ERA 2008]. They also estimate that direct energy consumption is less on a per unit product basis than embodied energies, which has been disproved by other others namely, Chapman and Roberts.

When compared to available data from mines operating at lower ore grades (Olympic Dam, Rossing), Equation 2.4 over predicts energy consumption by a factor of ten or more. Indeed, Storm van Leewen's model predicts that the Rossing mine (G = 0.03% U_3O_8) should consume twenty to fifty times more energy than was actually reported, depending on whether soft or hard ore is assumed. Even allowing for embodied energy, large disagreement from operational data is evident.

Again in the literature the necessity to look at the effect of ore grade on energy dependence was seen in [Rosa 2008] which noted that the average grade of copper ore mined in the USA over the last century has declined drastically while production has grown. Rosa realized that this reduction in grade had a direct effect on the energy consumption in the production of copper as he stated, "More resources are available at lower grades and at less accessible deposits, but then mining and processing energy needs become higher and larger amounts of overburden and wastes impact more heavily on the environment". [Rosa 2008] refers to a paper [Ruth 1995] considering the importance of not only the decline in average ore grades, but also the effects that technological improvements have in energy consumption of mining and milling. Rosa agreed with Smith and Storm van Leeuwen that the yields of mines show that the metal recovery rate decreases quickly at low ore grades while referring to equation 2.4. Rosa states that for uranium, “the yield drops to below 70% for grades smaller than 0.01%”. Rosa brought into light the importance in defining a peak ore grade for uranium, in which the same amount of energy is used in mining and refining the uranium product as would be created in a nuclear power plant. Rosa refers again to Smith and Storm van Leeuwen by accepting their estimate of the cut-off grade being .05%, an estimate much higher than other authors. No notable changes were made to Chapman’s model by these authors.

[Prasser 2008] expanded upon the model developed by Chapman and included a new term. The model is

$$e = \frac{100}{GY}\left(\left(1+S\right)e_{\min e} + e_{mill}\right) + e_{refine} \tag{2.6}$$

Here, the e_{refine}, has no dependence on ore grade. Prasser included this term because he found that the inverse relationship between energy and ore grade did not appear to hold given recent mine-reported operating data. Data in the late 1990s started being published in company mining reports, mainly due to an impetus to include metrics relevant to atmospheric carbon dioxide and other environmental sustainability factors. Based upon this modern data, Prasser observed that the energy intensity of the entire production process (including concentration, recovery by elution, precipitation, and drying, has a dependence on the ore grade that is more complicated than simple inverse proportionality. Therefore, Prasser added a term to the energy intensity model that allows the energy consumption is also to be proportional to the amount of uranium product.

Prasser used mining data from only one mine, the Rossing mine in Africa, years 1999-2006. The coefficients as well as their statistical quality are shown in Table 2.5. The statistical quality suffers due to the data all being from the same mine.

Table 2.5. Regression coefficients reported in [Prasser 2008]

Coefficient	Value	Statistical Uncertainty (1 Std. Dev.)
e_{mine}	0.023 GJ(t+e)/t ore + overburden	+/- 0.04 GJ(t+e)/t
e_{mill}	0.045 GJ(t+e)/t ore	+/- 0.33 GJ(t+e)/t
e_{refine}	52 GJ(t+e)/tU	+/- 123 GJ(t+e)/t

The statistic for e_{refine} makes it impossible to state that energy consumption is related to this step. The ore grade at Rossing is low, G= .03% U_3O_8 , and the model therefore doesn't show that with higher ore grades the first two terms have lower mass throughputs and energy is more dependent on the refining step. Prasser defines peak ore grade to be the grade at which the input energy equals

10% of the energy produced at a nuclear power plant and finds the peak grade to be .0014%. Differing from Chapman, Prasser's model does not include embodied energy. Studies account differently for embodied energy of fuels, materials and chemical inputs. The varying accounting along with [Rotty 1975] implies that on-site energy consumption is dominant.

The conclusion from this literature review is that Prasser's model best captures the grade-to-energy trend over a wide range of grades reported at modern mines, while forecasting future trends at lower ore grades. However, Prasser did lack a sufficiently large enough data set to prove the certainty of his model and this was made evident by the uncertainties around his coefficients. It is also possible that he misunderstood the e_{refine} term. The most likely case is that this coefficient is describing more than it is being given credit by Prasser, and this will be pursued in Chapter 3, while renaming e_{refine} to e_u. Prasser's model takes into account varying stripping ratios, ore grades, and yield, all of which influence the amount of energy consumed. Figure 2.4 shows energy (GJ/kg u) with decreasing ore grade (%), employing both Prasser and Smith and Storm Leeuwen's models. Actual mine data discussed by the investigators are shown on the figure as points of reference.

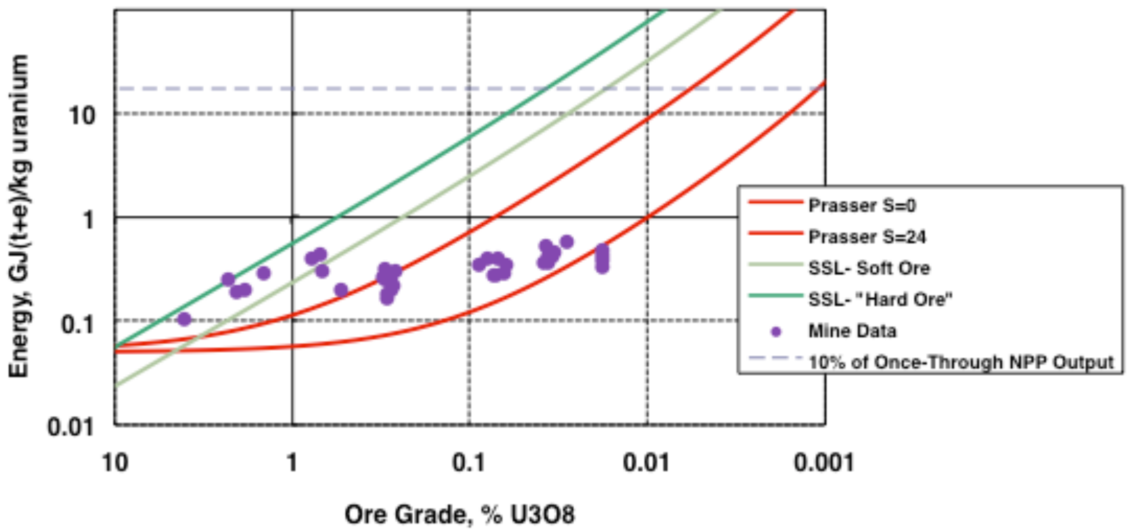

Figure 2.4. Summary of Literature Review

2.3.

To perform a top-down energy model for metal extraction, solid operating data was imperative. Annual numbers had to be collected for the three mass flows shown in Figure 2.1, as well as direct energy consumption. Historically, it has not been company standard practice to report energy consumption nor mass throughputs for their individual extraction projects. Mining companies may report annual totals for their entire company, but if not tied to individual deposits with known characteristics these are not useful to the present work.

Given rising concern over environmental greenhouse gas emissions in the last decade, however, it has become more common for companies to be detailed and specific in their reporting of energy use. When data is reported, it common practice for companies to report aggregated energy consumption, GJ(t+e), if energy is reported by carrier the numbers have been aggregated for this reports' purposes. Due to the crucial nature of proper data collection to the development of this report, this portion of the literature review is devoted specifically to the source data collected.

Much data was taken from company reports to stakeholders in the past decade. Although these annual reports often focus on net-earnings and future changes, they more recently have begun including mass flows specific to each mine. An example is shown from Rio Tinto's Rossing Report to Stakeholders of a uranium mine in Figure 2.5. This is typically the way these data are reported by companies. It is not shown in the figure, but annual average ore grade pertaining to each mine is often given in the body of these reports. When not specified, the less optimal approach is taken of dividing ore produced by ore mined to get % U_3O_8 in the ore. This requires that an approximation be made concerning mill efficiency (see chapter 3).

Performance data table	Target for 2009	Target for 2008	2008	2007	2006	2005	2004
Employees							
Number of employees	1,500	1,300	**1,307**	1,175	939	860	833
Production							
Uranium oxide produced (tonnes)	4,018	4,004	**4,108**	3,046	3,617	3,711	3,582
Ore processed ('000 tonnes)	13,253	13,133	**12,858**	12,613	12,008	12,027	10,972
Waste rock removed ('000 tonnes)	52,236	33,654	**33,899**	21,396	16,835	7,483	8,139

Figure 2.5. 2004-2008 Rossing Data

Also in the past decade, companies have begun to release sustainability reports for each of their mines. These reports often give energy use, sometimes by carrier, but more usually aggregated as one number. Figure 2.6 shows data from a mine-specific company sustainability report. In this case, both energy use and mass throughputs were given for this gold producing mine.

	2004	2005
Ore mined (tonnes)	2,766,033	3,043,595
Total material moved (tonnes)	17,049,525	14,400,476
Ore crushed (tonnes)	2,753,876	3,006,390
Average gold grade (g/t)	0.96	1.03
Gold produced (ounces)	76,186	62,471
Employees	113	100
Mine life	March-07	March-07

Energy Use	**2004**	**2005**
Electricity (MWh)	18,467	17,247
Diesel Fuel (kL)	7,052	5,869

Figure 2.6. 2005 Wharf Sustainability Report.

In another specific case data was found in an annual social and environmental report that was published by Energy Resources of Australia [ERA 2008]. This gave ten years of mine performance.

Gavin Mudd [Mudd 2007],[Mudd 2007a],[Mudd 2007c],[Mudd 2008a],[Mudd 2010] has a series of publications related to the mining and milling of uranium, gold, and copper, with energy and ore grade reported. Data was extracted, in some cases, from his papers as well as from his references. His data was most useful in the case of gold, in which difficulty was found in tracking down company reporting.

The data in Appendix A made this report possible and should be referred to in order to understand variation in ore grade among metals as well as the details of the complete data set. A review of the literature returned no direct comparison of energy consumption between uranium mining and the mining of other minerals. Therefore, this study expands on the reviewed work by assembling recent mine performance data. 1960s and 1970s data will not properly model 21st century mining, due to advancements in technology and the improvement in company reporting standards. More validity will be given to the uranium mine energy intensity model by applying it to data for three different minerals to show its applicability over a wide span of ore grades.

Chapter 3

Methodology

This section first describes the model used for defining the energy-to-grade relationship for uranium mines. The reasoning for comparing different commodities using the same model follows. Next the model will be broken down to discuss the interpretation of each coefficient. Lastly, the collected data will be presented numerically while graphically and qualitatively discussed.

3.1

Prasser's model for energy consumption is based on the mass flows through the mining and milling processes and his model is hereby adopted. It was seen in the literature review that his model best represented the energy-to-grade relationship of a full range of grades at different mines. This top-down model is used to forecast energy consumption as well as depict declining ore grades among certain commodities. While a bottom up model would be ideal for studying a single mine, a top-down model is appropriate for long-term forecasting and studying a range of mines. A top-down model uses a mathematical expression to represent the quantities dependent upon normalized energy consumption [GJ/ t product]. For metal extraction these quantities are ore grade, mill and/or leaching process yield, and the mass of waste rock relative to the mass of ore.

Prasser included a coefficient to his model that had no dependence on ore grade, and the justification of this will be explored in this section. The e_{refine} coefficient added by Prasser is now renamed $e_{product}$, because it is thought that the coefficient accounts for more than Prasser gave it credit for in his model. This model asserts that energy consumption is based on mining method and the three mass flows, mining, milling, and refining. The energy intensities for each will be calculated from the mining data in Appendix A using a linear regression analysis. The regression coefficients will then be applied directly to the model equation along with a wide range of ore grades (.002-20%) and associated yields based on equation 2.5, for uranium. This will show how energy consumption varies with ore grade using regression coefficients based on actual data. The coefficients can be compared to the mass-flow based energy coefficients from past studies for evaluation purposes. In addition, the coefficients can be applied directly to individual mine data to predict energy consumption at the mine. For example, multiplying the coefficients by their corresponding annual throughput and applying the mines average ore grade to 3.1, results in the energy consumption at that mine. This can be compared directly to actual mine reported energy consumption to show the goodness of fit of the regression analysis.

The expression for both open pit and underground mining, the terms of which are described in table 2.2, is as follows,

$$e = \frac{100}{GY}\left(\left(1+S\right)\left(e_{\min e,OP} + e_{\min e,UG-OP}\delta_{UG}\right) + e_{mill}\right) + e_{product} \tag{3.1}$$

if it is an underground mine, δ_{UG} =1

if it is an open pit mine, δ_{UG} =0

where $e_{mine(UG\text{-}OP)}$ represents the difference in energy intensity between underground and open pit mining. The delta function allows for both open pit and underground mines to be modeled in a single regression, with the same milling and product refining energy intensity coefficients but allowing for different mining intensity coefficients.

The data for uranium underground mines was limited, and had consistently low stripping ratios as expected for this mining technique. Due to the limited range of stripping ratios represented by the mines in the uranium data set, the regression analysis had difficulty in differentiating the energy consumption between mining and milling. This led to poor statistical confidence in these two parameters and it was necessary, for uranium, to aggregate mining and milling, e_{mm}, for underground mining:

$$e = \frac{100}{GY} e_{mm} + e_{product} \qquad \text{(underground uranium mining)} \qquad (3.2)$$

It should be noted that e_{mm} is the energy (GJ(t+e) per tonne ore) to mill one tonne of ore plus the energy to mine the overburden with low stripping ratios comparable to the mines used in the regression analysis ($S < 1$). While this condition limits the scope of the underground model, the restriction is not as limiting as it may seem. Stripping ratios in underground metal mines are generally considerably lower than in open pit mines [IAEA 2000].

In addition to regressing the Appendix A data, it was also beneficial to regress the data after dropping the product term to observe the change in energy consumption. While there is an energy consumption associated with $e_{product}$, it can be seen in equation 3.1 that this coefficient has no grade dependence. To better understand the role that $e_{product}$ plays in describing energy consumption, It was therefore beneficial also to explore the predictive power of the model in the absence of the $e_{product}$ term. In this case, equation 3.1 becomes,

$$e = \frac{100}{GY}\left((1+S)(e_{\min e,OP} + e_{\min e,UG-OP}\delta_{UG}) + e_{mill}\right) \qquad (3.3)$$

and the same methods of evaluating the model as previously discussed are used. The model should predict a linear (1/G) relationship proportional to energy consumption for low ore grades. However, if the trends observed for uranium extend to other commodities the model will be seen to fail to predict energy consumption at higher ore grades.

There are standard errors associated with each coefficient and these errors can be propagated over equation 3.1 and 3.3 to create error bars. Equation 3.4 shows he model for the error bars. When applied to equation 3.1 the sigma product term is withheld. The sigma term for mining can be varied according to what type of mining is under consideration.

$$\sigma_E = \sqrt{\sigma_{product}^2 + \sigma_{\min e}^2 \left(\frac{S}{GY}\right)^2 + \sigma_{mill}^2 \left(\frac{1}{GY}\right)^2} \qquad (3.4)$$

3.2

Since the processes of metal extraction and refining are analogous among metals, it was useful to compare the model that was developed for uranium to other commodities. It should, however, be noted that not all metals are extracted or refined the same way, but the metals considered hereafter have a close enough analogue to base these comparisons [Chapman 1983]. The model was applied to a metal with higher and a metal with lower crustal abundance than uranium, see Table 3.1.

Table 3.1 Crustal abundance of metals used, Source: [Wikipedia 2010]

	Symbol	**Avg. Crustal abundance (ppm)**
Gold	Au	.004
Uranium	U	2.8
Copper	Cu	60

Equation 3.1 was used model all data collected for gold and copper, and included data from both underground and open pit mines. When mining technique was not reported at the mine an assumption was made that grades above ~5 ppm were underground mines [Mudd 2007c]. More data was collected for the underground mining of gold, than uranium, and could be directly modeled by the equation 3.1, because of the variation in stripping ratio. It was found that

stripping ratios for underground gold mining are larger than that of uranium, possibly due to the lower crustal abundance and similarly lower ore grades. Due to lack of useful mining data, only open pit mining will be considered for copper. Because of the similarities among commodities reported in the literature, it is expected that the metals will follow a similar energy-to-grade relationship as is expected for uranium. The formula for yield that was introduced for uranium in Chapter 2, equation 2.5, of this report cannot be applied to copper and gold as it was made, by [Storm 2007], specifically for uranium. The yield is site specific while dependent on varying technologies and equipment, and it is impossible to know the exact yield for each project. However, companies will want to keep their yield as high as possible and for the purposes of this report an optimistic yield of 0.98 will be used for gold and copper, while equation 2.5 will be used for uranium.

The data sets used to define energy use can be found in Appendix A of this report. It was necessary, in some cases, to manipulate the data to have a uniform data set. This manipulation was done be aggregating energy consumption from thermal and electric carriers were to GJ(t+e).

For example, when electrical energy is reported in units MWh(e) and thermal energy as liters(L) of diesel fuel, the following is used to develop uniform consumption units: [Supple 2010]

$$GJ_{(e+t)} = 3.6(MWh_e + 0.0358(L)$$
$$MWh_e = 3.6GJ_{(e+t)}; L(diesel) = 0.0358GJ_{(e+t)}$$

The other manipulation made to data was for the case of the copper co-producing mines and the Olympic Dam mine, metrics (energy, waste, throughput) are calculated based on their revenue, derived from the amount of product. Companies reported the metrics as totals for their co-producing mines. Following the strategy of [Mudd 2010], for the Olympic Dam underground co-producing mine in which 20% of total reported metrics were assigned to uranium, we disaggregated the minerals' metrics based on their annual value. Revenues were used from the US Geological Survey [USGS 2010] in dollars per tonne to allocate the metrics based on their annual worth.

The strategy for copper is as follows:

where,

X_i =tonnes (any metal) , X_1 =tonnes copper, and $X_{i\$}$ =annual revenue ($) from metal

M_{total} =metric total for mine, Mx_i = metric for copper

$$X_{i\$} = \frac{S}{X_i}(X_i)$$

$$\%X_{1\$} = \frac{X_{1\$}}{X_{1\$} + X_{i\$}}$$

$$M_{x_1} = \%X_1(M_{total})$$

The metrics, M, are either GJ, waste rock, or ore milled.

Using this method, at two of the copper mines the reported energy accounted for 90% of the energy based on copper's revenue against the other metal in production at the mine. In the third copper-nickel mine, copper accounted for 20% of the total energy consumption, because nickel is worth an order magnitude more than copper. It is true the inclusion of co-product mines is not ideal and introduces some double accounting, but limited data made this necessary.

Mining data in Appendix A are broken up into companies and labeled as open pit, underground, and/or co-producing. There are a total of four copper, eight uranium, and 30 gold mines. All of the copper mines are open pit and several are co-producing. A single uranium mine and eight of the gold mines are underground. It should also be noted that the gold product has units of ounces, whereas copper and uranium are reported in tones; this notation is commonly found in the mining industry. Finally, for all mines, energy in the tables refers to direct energy use only. As described in Chapter 2, the energy pertaining to the consumption of chemicals, machinery, and machine parts (embodied energy) is not included.

Figures 3.1, 3.2, and 3.3, give energy on per product basis versus ore grade for the data sets used in this report. These figures do not show the grade-to-energy relationship strictly following the (1/G) behavior over all ore grades, as postulated by the models in Chapter 2. As described in the previous chapter it has been understood that the inverse grade-to-energy relationship would be expected to be apparent at lower ore grades, establishing a cut-off grade for any metal.

The available data for uranium and gold spans several orders of magnitude in ore grade for each metal. The data points indicate that grade alone cannot explain the observed energy consumption trends. Evidently a substantial component of the energy consumption, especially at higher-grade mines, is not tied to the ore grade. It is most likely that this non-grade dependent consumption is tied to the refining step, where process mass flows are proportional to the product mass. This energy consumption may also be tied to inherent energies such as transportation from the mine to the mill. Although most mills are collocated with the mines, there is still transport involved and may be a hidden consumption that has no dependence on the grade of the ore. However, it is difficult to know exactly what is behind this non-grade dependent energy consumption. This is behind the reasoning for including the coefficient with no grade dependence, $e_{product}$. This is also why the model is evaluated without this coefficient, in order to show its' importance. The same is true for the figure 3.3, with copper data, however it is more difficult to draw a conclusion in this case because there are so few data points.

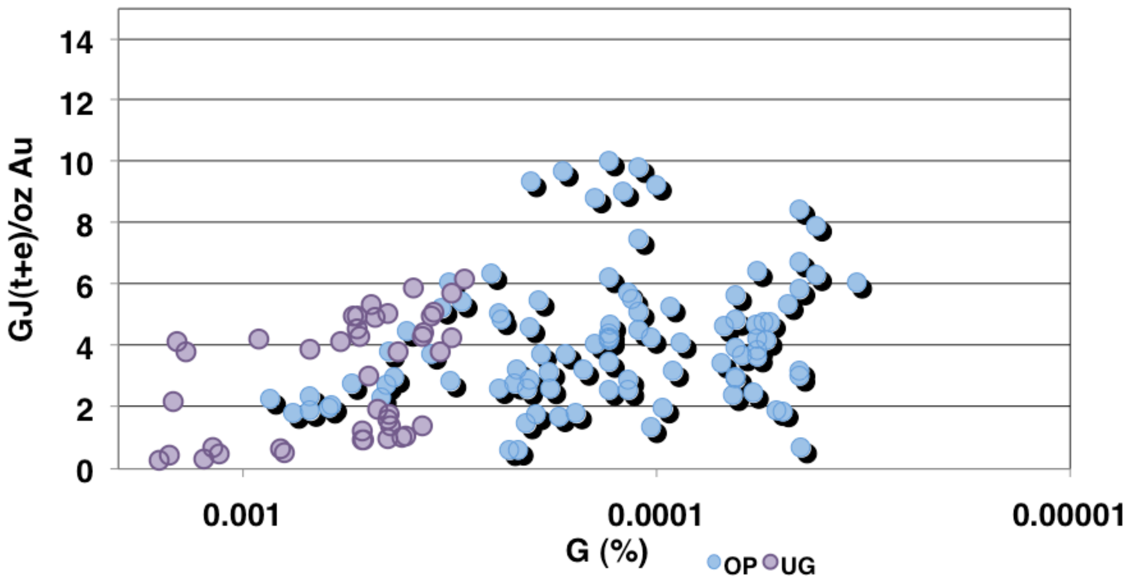

Figure 3.1 Energy consumption versus ore grade at gold mines, data from Appendix A. (open pit (OP) and underground (UG) mining)

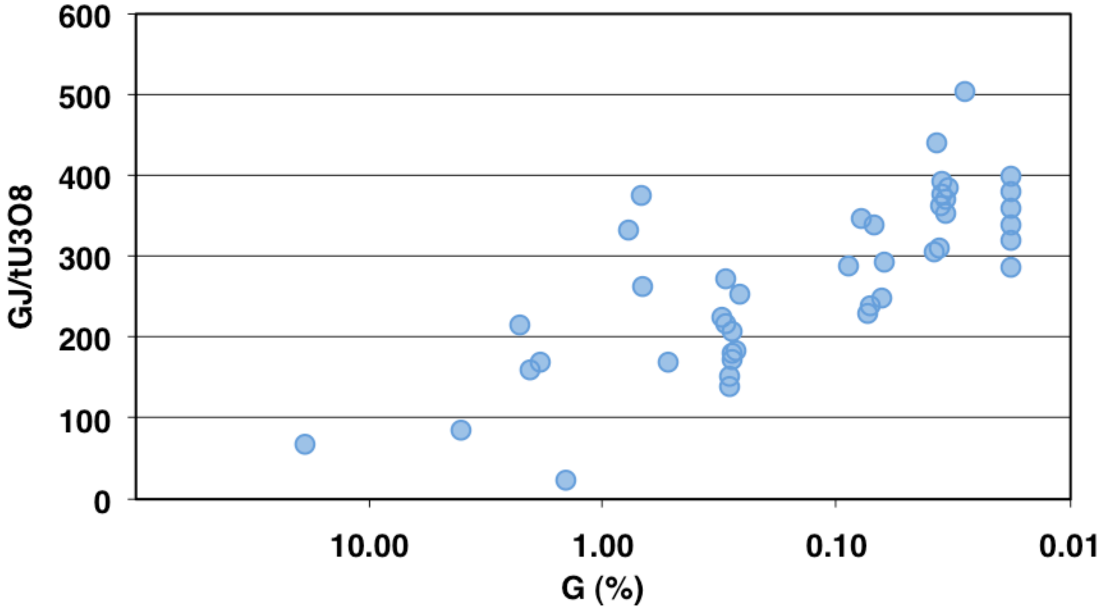

Figure 3.2. Energy consumption versus ore grade at uranium mines, data from Appendix A.

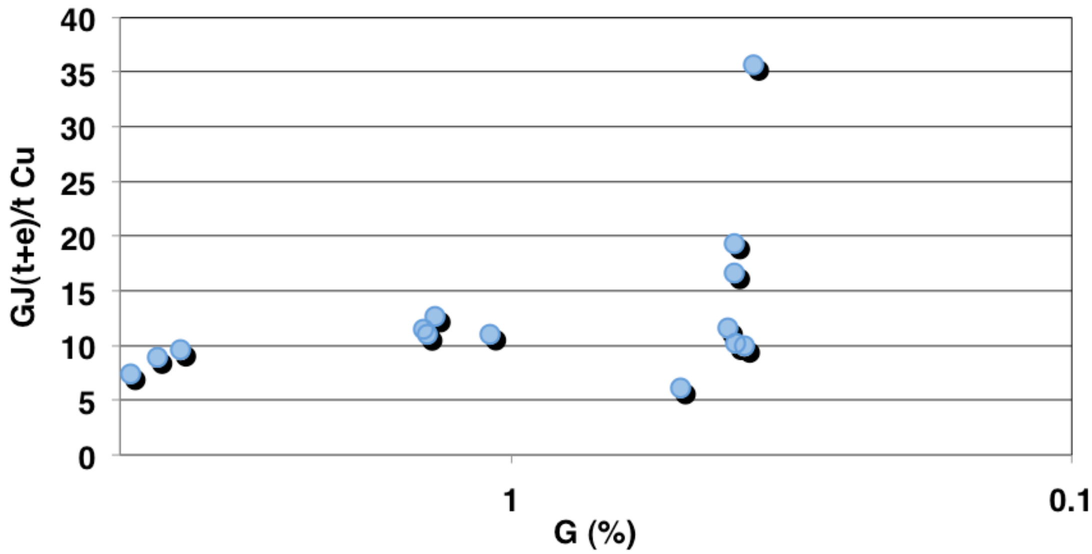

Figure 3.3 Energy consumption versus ore grade at copper mines, data from Appendix A.

The methods of this report have been developed from that of the investigators described in Chapter 2. A large data set, with varying ore grades, was compiled to define and forecast energy consumption. These data are used to evaluate the top down model presented earlier. In the next chapter, the regression results will be presented and the ability of this model to describe the commodities of gold and copper will be examined. Finally, the consequences of dropping the constant term, $e_{product,}$ from the model, will be explored.

Chapter 4

Results

This chapter presents the results of the uranium, gold, and copper regression analyses and draws conclusions regarding the applicability of the energy intensities to the other commodities. The data that the results are based upon is available in Appendix A of this report.

4.1 Uranium

Table 4.1 shows the coefficients obtained for open pit (OP) and underground (UG) mining from regressing the uranium mine data from Appendix A, for the 28 OP and 7 UG data points, onto equations 3.1 and 3.2 using the statistical analysis toolkit in Microsoft Excel. The R-squared value for the fit was 0.904. The statistical quality associated with this fit is acceptable, with the exception of the coefficient for mining e_{mine} where the t-statistic is low. This is where the regression falls short due to lack of data from mines operating at high stripping ratios.

Table 4.1. Energy intensity coefficients obtained from regression analysis.

Coefficient	Applies to	Value	Standard Error [GJ(t+e)/t]	T Statistic
$e_{product}$	OP, UG	178[GJ(t+e)/tU]	12.2	14.6
e_{mill}	OP	0.0236[GJ(t+e)/(t ore)]	0.0053	4.44
e_{mine}	OP	0.0125[GJ(t+e)/t(ore+ob)]	0.0119	1.04
e_{mine}	UG	0.291[GJ(t+e)/(t ore)]	0.0340	8.55

The current model improves upon Prasser's predictions for ore grades of greater than 0.1% U_3O_8 where the $e_{product}$ term governing uranium refining and other grade-insensitive energy inputs plays a much larger role in determining energy use. The current model agrees with Prasser at lower grades where ore milling and overburden haulage dominate the energy balance. Error bars are shown on Figure 4.1 for the current model.

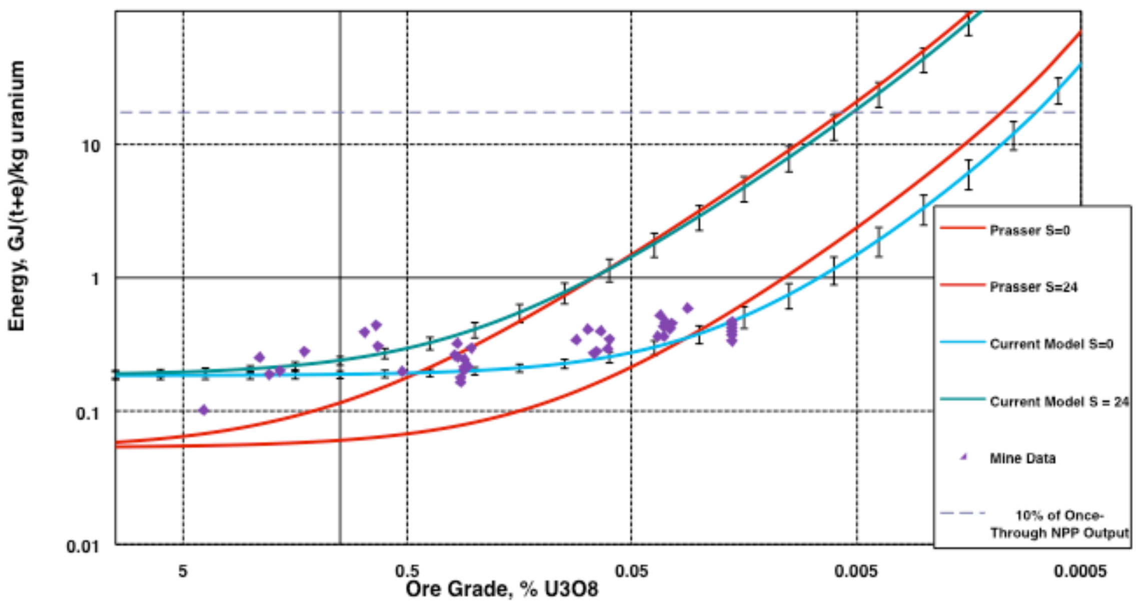

Figure 4.1. Results of Prasser and current model

Figure 4.2 shows actual mine report energy consumption verse energy consumption predicted by the intensity coefficients from the regression. The figure shows the results of the Rossing mine in Namibia, Africa, whose average ore grade is .03%.

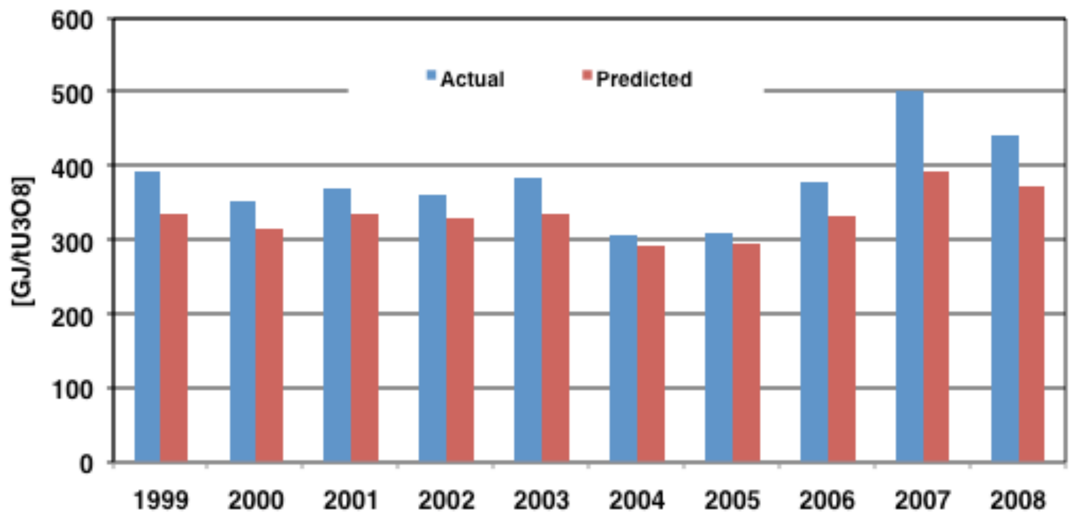

Figure 4.2. Energy Consumption, actual versus predicted, data: Rossing uranium mine in Appendix A.

4.2 Gold

The coefficients are presented in Table 4.2 after the gold data was regressed on equation 3.1 using the 98 OP and UG 42 mining data points. The large data set benefitted the regression analysis and gave rise to high t-statistics, but the range of ore grades, specifically 0.3-16 ppm, was more limited than that seen for the operating uranium mines. In Table 4.3 the coefficient values are given for e_{mine} and e_{mill}, while the product term has been dropped. The coefficients have the same order of magnitude as the values when the constant is dropped, indicating that for the deposits and ore grades currently being tapped for gold recovery, a strict (1/G) model might describe energy consumption rather well.

Table 4.2 Energy intensity coefficients obtained from regression analysis

Coefficient	Applies to	Value	Standard Error [GJ(t+e)/oz or t]	T Statistic
$e_{product}$	OP, UG	0.77[GJ(t+e)/oz Au]	0.2	2.6
e_{mill}	OP, UG	0.0117 [GJ(t+e)/(t ore)]	0.0046	4.2
$e_{mine,OP}$	OP	0.02913[GJ(t+e)/t(ore+ob)]	0.0032	8.9
e_{mine_UG}	UG	0.0071[GJ(t+e)/t(ore+ob)]	0.0046	7.8

Table 4.3. Energy intensity coefficients obtained from regression analysis, no constant

Coefficient	Applies to	Value	Standard Error [GJ(t+e)/oz or t]	T Statistic
e_{mill}	OP, UG	0.0114 [GJ(t+e)/(t ore)]	0.0039	7.6
$e_{mine,OP}$	OP	0.037[GJ(t+e)/t(ore+ob)]	0.0032	9.2
e_{mine_UG}	UG	0.00059[GJ(t+e)/t(ore+ob)]	0.0045	8.07

On the other hand, it should be noticed that the model provides poor predictions for higher-grade deposits when the grade-independent term is dropped. Figure 4.3 shows the importance of having a product term included in the model. At larger ore grades this term evidently dominates the energy consumption directly associated with mining operations. In figure 4.3, the underground gold mine regression function was evaluated at a stripping ratio S=2.2.

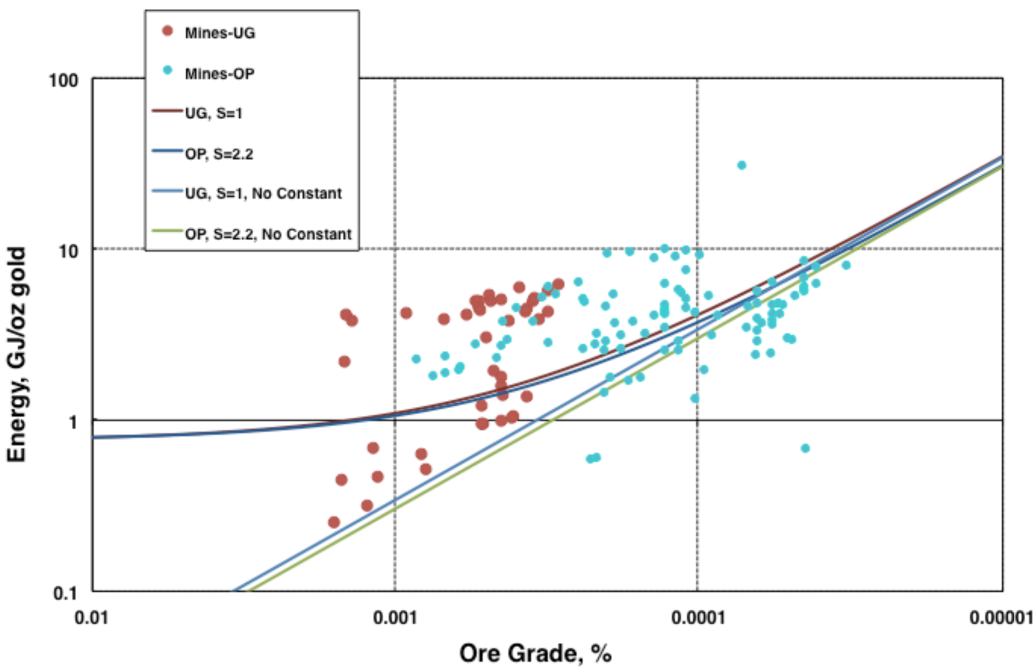

Figure 4.3. Results of current model

Figures 4.4 and 4.5 include the standard error after being propagated on the model equation for open pit and underground, respectively.

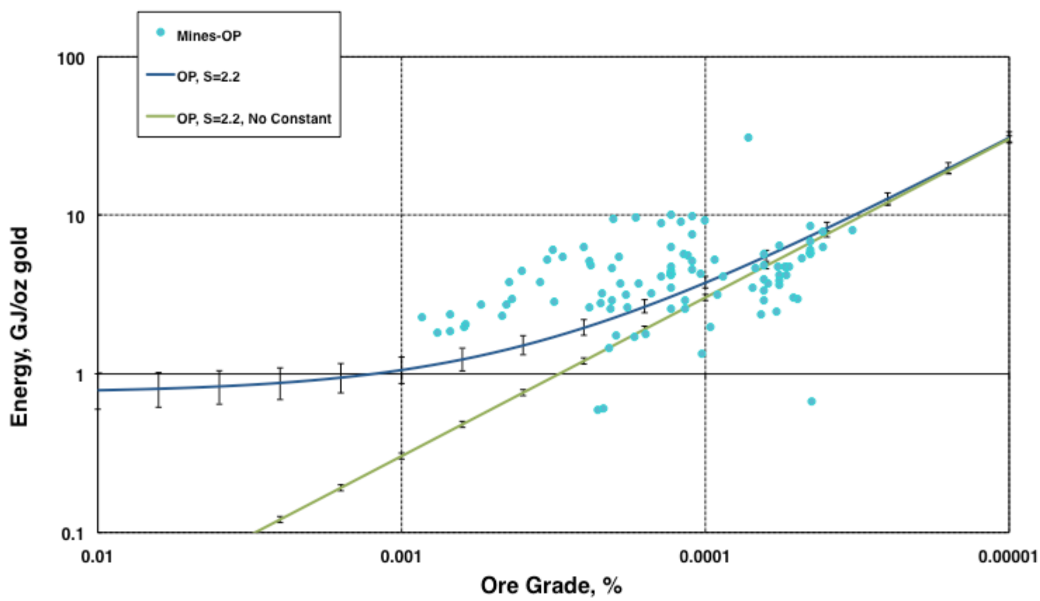

Figure 4.4. Results of current model, Open Pit

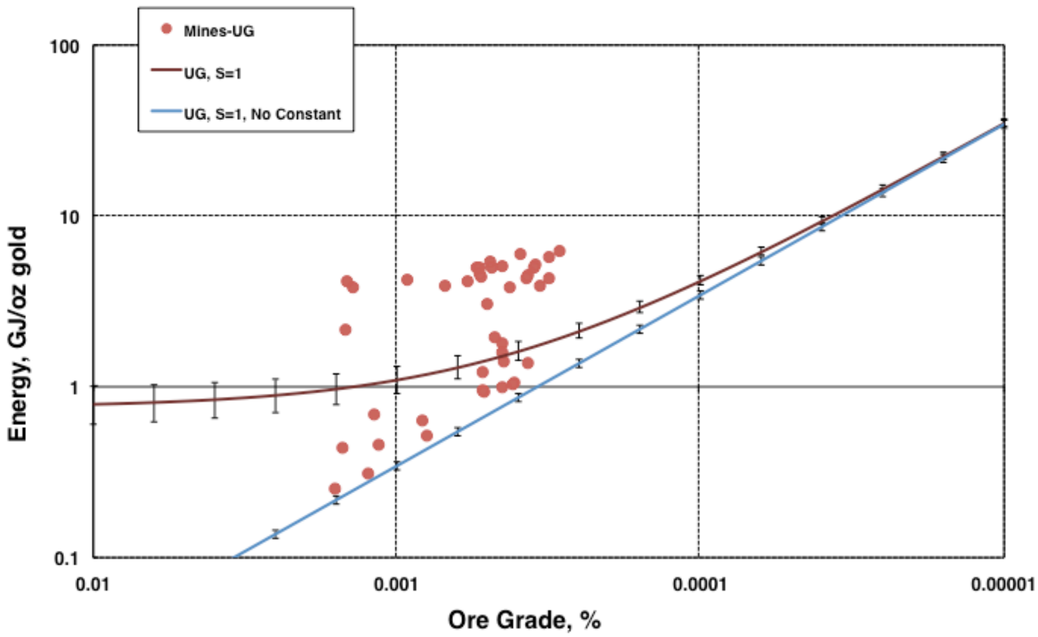

Figure 4.5. Results of current model, Underground

Figure 4.6 and Figure 4.7 apply the model to specific mines. In this case the coefficients were applied to three open pit mines and four underground mines, respectively. The figures show actual company reporting direct energy and energy consumption predicted by the regression coefficients.

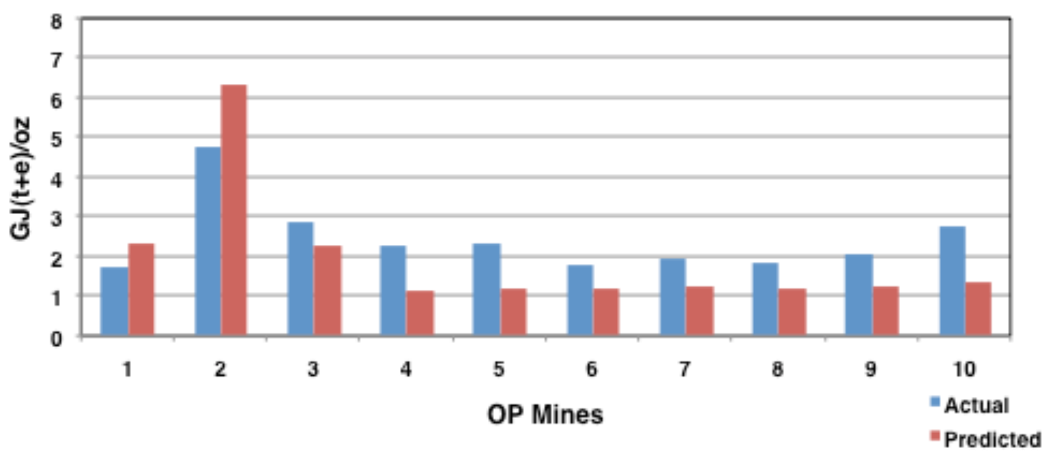

Figure 4.6. Energy Consumption, actual versus predicted, data:(from left to right) Bald Mountain (2006-2007), Ruby Hill(2007), and GoldStrike (2001-2007) open pit gold mines available in Appendix A.

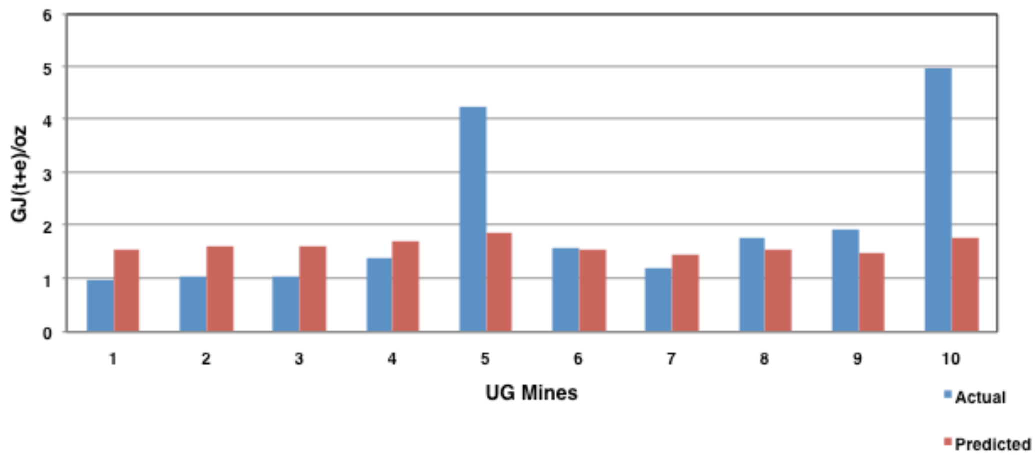

Figure 4.7. Energy Consumption, actual versus predicted, data:(from left to right) Hemio (2004-2007), Plutonic(2001), Lawlers(2004-2007), and Sunrise Dam(2002-2003) underground gold mines available in Appendix A.

Figures 4.6 and 4.7 depict that in each case shown energy per unit product is the same order of magnitude for both actual and predicted consumption.

The results from this analysis produce comparable results to that of [Mudd 2007c] for gold, which got 0.85-7.8 in GJ/oz product. The current result, which also combined open pit and underground, was 0.90- 10 GJ/oz product.

4.3 Copper

Regression coefficients for copper are given in Table 4.4 and Table 4.5, using 17 open pit mining data points. Statistical measures indicated that the model provided a poor fit. This is most likely attributed to the small data set and small variation in ore grade for this commodity. It is also probable that the uncertainty is associated with three of the four mines being co-producing mines (to account co-products a revenue based disaggregation described in chapter 2 was used, but it is impossible to know the true 'share' of energy consumption associated with copper at co-producing mines). This is apparent in the statistics from the regression analysis, as we see the t-statistics suffer for each coefficient but especially for the product, where the null hypothesis cannot be rejected.

Table 4.4 Energy intensity coefficients obtained from regression analysis

Coefficient	Applies to	Value	Standard Error [GJ(t+e)/t]	T Statistic
$e_{product}$	OP, UG	2.45[GJ(t+e)/t Cu]	2.6	0.91
e_{mill}	OP, UG	0.0484 [GJ(t+e)/(t ore)]	0.0203	3.2
e_{mine}	OP, UG	0.0151[GJ(t+e)/t(ore+ob)]	0.0061	2.4

When the product term is removed the statistics still suffer and although the t statistic seems reasonable, the error for the milling term is four times that of the true value.

Table 4.5. Energy intensity coefficients obtained from regression analysis, no constant

Coefficient	Applies to	Value	Standard Error [GJ(t+e)/t]	T Statistic
e_{mill}	OP, UG	0.0075 [GJ(t+e)/(t ore)]	0.029	1.2
$e_{mine,OP}$	OP	0.040 [GJ(t+e)/t(ore+ob)]	0.012	3.3

Figure 4.8 graphically depicts the regression results. Again it is seen when the constant is dropped from equation 3.1, the model does not describe high ore grades. The significance of the large standard error is shown here through the error bars on the model. It is difficult to conclude the effect of removing the product term from the model here because the statistical uncertainties associated with the two models are seen to overlap.

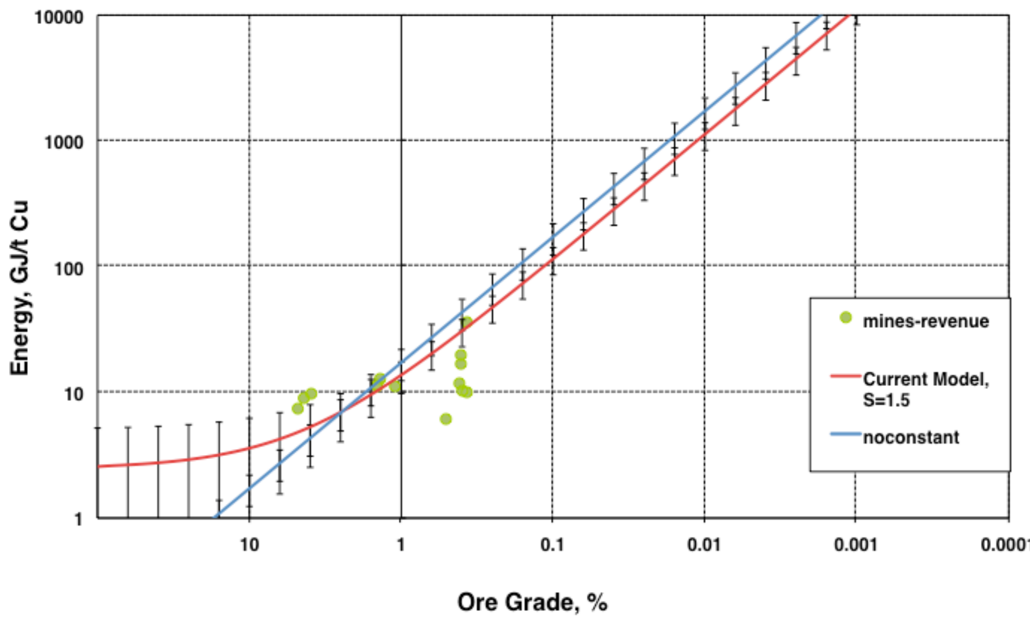

Figure 4.8. Results of current model

All of the copper mines that were used for the analysis are seen in Figure 4.9, where the predicted energy consumption for almost half of the mines are twice as much as the actual reported energy consumption. Clearly, more data is needed to confirm the models' relevance to copper. It would be ideal to model a commodity, of higher ore grade, that is not co-produced, because of the difficulty in apportioning metrics to each commodity at the mine/mill.

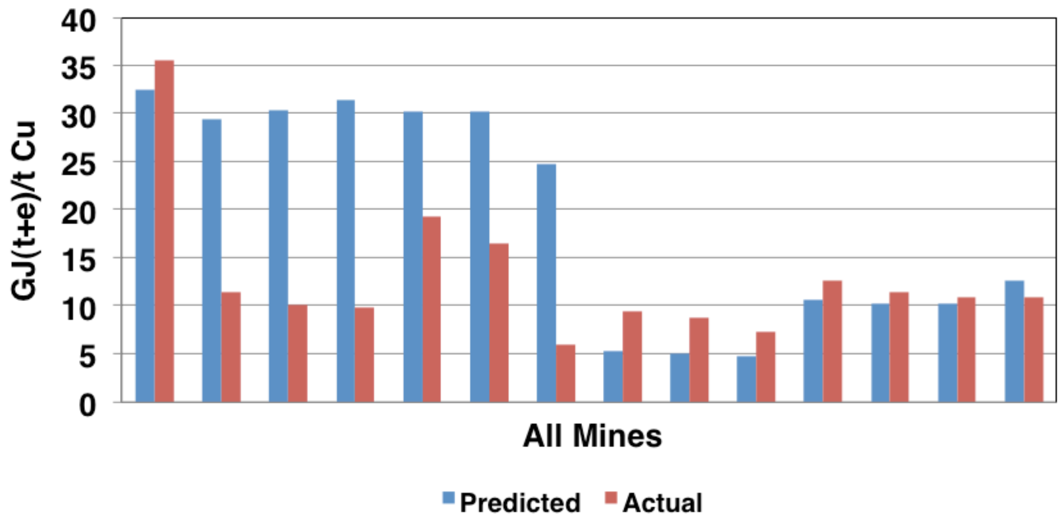

Figure 4.9. Energy Consumption, actual versus predicted, data: all copper mines in Appendix A.

Chapter 5

Conclusion

This report was motivated by from the perceived uncertainties in the energy consumption of steps in the front-end nuclear fuel cycle. Large discrepancies were found in the magnitude of energy intensity in mining and milling of uranium. The goal here was to model how and where energy was being used in these processes while confirming the dependence of energy use on ore grade. In order to validate the model for uranium, it was beneficial to compare it to other metals of higher and lower ore grades. Copper and gold were chosen because of their crustal abundance as well as their seemingly adequate data sets. Energy consumption in the mining and milling of three different commodities has been presented and analyzed.

In conclusion of this report, the model presented has confirmed that energy consumption will increase as ore grade decreases for uranium ore, but that a strict inverse-grade model cannot accurately depict energy use at both high- and low-grade mines. By using far more data than any investigator had previously, the quality of the statistics were increased. The model depicted the energy-to-grade relationship for gold with good statistical accuracy. The reasonable results for modeling gold concluded that the model described for uranium could be applied to other commodities of different crustal abundances. However, it was found that the data set acquired for copper was not sufficient to describe the energy-to-grade relationship for the commodity. Unfortunately copper is usually co-mined and produced, making it difficult to apportion energy values to the individual extracted commodities. Due to this ascription inaccuracy, it is difficult to draw concrete conclusions on whether or not the model presented for uranium and gold can accurately describe copper as well. In continuing upon this research, it would be valuable to choose another, higher-abundant, commodity to use as a comparison to the model. In addition, the constant term, $e_{product}$, should be addressed. It was seen when the constant term for gold was dropped from the equation that the current data was reasonably explained by a strict (1/G) model. However, the higher-grade ores were better represented when the constant was kept in the equation but the statistics did not support definitive conclusions regarding the utility of this term. It seems more beneficial at this stage to keep the constant term in the equation in order to describe that energy consumption for $e_{product}$ that does not have grade dependence. The conclusion should be re-drawn when the data set has more variation in ore grade.

Lastly, this report can inform the nuclear industry at large. Looking forward in the 21st century, there will be more and more concern placed on energy consumption due to its strict relationship with carbon dioxide emissions. While a new nuclear age is currently being defined, it is necessary for the industry to competitively explore and explain ways in which nuclear compares and exceeds other base-load powers. This study has taken an in-depth empirical approach to quantifying energy consumption in a critical step of the fuel cycle. Now energy consumption can be modeled for future mine generations as ore grade slowly declines, allowing meaningful comparison to the energy consumption of other commodities and fuel cycles.

Appendix A

Complete Listing of Uranium Mine and Mill Data

Table A.1. Rio Tinto (Open Pit)

Year	Energy Consumed (TJ)	Waste(kt)	Ore Milled (kt)	U3O8 (t)	Ore Grade (ppm U)
Rossing					
1999	1248	15607	10463	3171	357
2000	1133	9787	11039	3200	341
2001	979	12033	9084	2643	342
2002	999	13015	8969	2752	361
2003	915	10434	8347	2374	335
2004	1096	8139	10972	3582	384
2005	1152	7483	12027	3711	363
2006	1366	16835	12008	3617	354
2007	1534	21396	12613	3046	284
2008	1812	33899	12858	4108	376

Source:

Annual Report to Stakeholders, Rössing Uranium Ltd, Years 1999 to 2009, Swakopmund, Namibia, available: www.rossing.com, webpage accessed January 5, 2010.

Table A.2. Energy Resources of Austraila (Open Pit)

Year	Energy Consumed (TJ)	Waste(kt)	Ore Milled (kt)	U3O8 (t)	Ore Grade (ppm U)
Ranger					
1997	749	5871		4162	
1998	906	5343			
1999	808	4524	1827	4375	2700
2000	922	5835	1550	4244	3000
2001	916	3485	1510	6564	2900
2002	810	2819	1784	4470	2800
2003	873	4249	2068	5065	2800
2004	1064	8500	2086	5137	2800
2005	902	14910	2293	5910	2900
2006	1205	9900	2072	4748	2600
2007	1223		2900	5412	3100
2008	1457		3500	5339	3000

Sources:

Mudd, G.M., The Sustainability of Mining in Australia: Key Production Trends and Their Environmental Implications for the Future, Research Report No RR5, Department of Civil Engineering, Monash University and Mineral Policy Institute, October 2007.

Annual Social and Environment Report, Energy Resources of Australia Ltd (ERA), Years 2001 to 2008, Sydney, NSW, available: www.energyres.com.au, webpage accessed January 2, 2010.

Table A.3. BHP Billiton (Underground)

Year	Energy Consumed (TJ)	Waste(kt)	Ore Milled (kt)	U3O8 (t)	Ore Grade (ppm U)
Olympic Dam					
1998	604	54	681	1740	790
1999	924	108	1349	3198	890
2000	1037	142	1780	4500	740
2001	1043	149	1867	4355	720
2002	976	142	1775	2881	690
2003	933	134	1677	3176	630
2004	1089	142	1777	4370	640
2005		154	1929	4362	620
2006		145	1817		570

Source:

Mudd, G.M., The Sustainability of Mining in Australia: Key Production Trends and Their Environmental Implications for the Future, Research Report No RR5, Department of Civil Engineering, Monash University and Mineral Policy Institute, October 2007.

Table A.5. Cameco/Areva (Open Pit)

Year	Energy Consumed (TJ)	Waste(kt)	Ore Milled (kt)	U3O8 (t)	Ore Grade (ppm U)
McLean Lake					
2002	594	140	122	2761	22900
2003	437	152	132	2733	20700
2004	456	170	148	2681	18600
2005	588	199	173	2451	14500
2006	214	150	131	814	6800
2007	146	196	170	864	5300

Source:

Mudd, G.M., M. Diesendorf, "Sustainability Aspects of Uranium Mining: Toward Accurate Accounting?" *2nd International Conference on Sustainability Engineering & Science*, Auckland, New Zealand, 20- 23 February 2007.

Table A.6. Data for other mines (Open Pit)

Year	Energy Consumed (TJ)	Waste(kt)	Ore Milled (kt)	U3O8 (t)	Ore Grade (ppm U)
McArthur River					
2007	586	51	44	8489	190000
Key Lake					
2006		252	219	8462	39100
2007	734	244	212	8483	40700
Rabbit Lake					
2006	787	360	313	2359	7800
2007	686	314	273	1825	6900
Cluff Lake					
2002		82	72	1917	27000

Source:

Mudd, G.M., M. Diesendorf, "Sustainability Aspects of Uranium Mining: Toward Accurate Accounting?" *2nd International Conference on Sustainability Engineering & Science*, Auckland, New Zealand, 20- 23 February 2007.

Complete Listing of Gold Mine and Mill Data

Table A.7a. Barrick Gold Data (open pit)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Bald Mountain						
2006	535259	12232	6475	304971	1.95	2
2007	641714	17894	8688	134933	0.55	5
Ruby Hill						
2007	484264	18704	3460	168941	2.02	3
Goldstrike						
2001	5612277	131584	9582	2482552	8.57	2
2002	5270220	120270	10850	2248887	6.86	2
2003	4171614	119443	10581	2315806	7.54	2
2004	4199887	107912	11199	2131507	6.17	2
2005	4139882	112674	10510	2220365	6.86	2
2006	4173190	109514	10825	2045939	6.10	2
2007	4901551	114601	10745	1787043	5.45	3
Round Mountain						
2001	2008671	10508	53216	819473	0.58	2.451
2002	1969714	838	56448	828249	0.65	2.378
2009	1205870	20040	30035	213946	0.64	5.636
2008	1193820	20658	37368	246946	0.64	4.83
2007	1189813	23717	36990	302971	0.64	3.927
2006	1115748	19014	43436	335115	0.64	3.329
2005	1085723	5807	61696	373115	0.64	2.910

Table A.7b. Barrick Gold Data (open pit)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Granny Smith						
1995	1311739	29099	4034	240311	1.92	5.458
1996	1484974	34138	3902	293250	2.40	5.064
1997	1452635	32649	4200	324464	4.00	4.477
1998	1399178	12128	4166	604670	4.63	2.314
1999	1570049	22953	4138	573778	4.50	2.736
2000	1687012	27800	4058	451975	3.50	3.733
2001	1980839	18408	3633	380817	3.30	5.202
2002	1772000	24677	4175	597615	4.30	2.965
2003	1943743	25448	3955	307273	2.50	6.326
1998	2044835	25761	5103	203930	1.30	10.0
1999	2132561	32139	6243	217325	1.10	9.813
2000	2156785	23432	6086	238863	1.20	9.029
2001	1688852	2786	5738	182746	1.00	9.2
Pierna						
2004	1161122	21300	15192	708595	1.028	1.335
2005	1387899	28050	14483	688851	1.714	1.69
2006	1847338	37426	15457	558321	1.166	2.894
2007	1671368	28232	16209	559417	1.166	2.547

Table A.7.c Barrick Gold Data (open pit)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Launas Norte						
2005	985854	8513	12945	603293	2.057	1.46
2006	891847	5367	19448	1189036	2.263	0.588
2007	935326	3385	19692	1191230	2.160	0.604
Kidston						
1995	1056873	20907	6103	225869	1.29	4.679
1996	1056244	31841	6093	191481	1.14	5.52
1997	1199461	39750	6150	209821	1.17	5.717
1998	1052466	24126	6845	206259	1.10	5.103
1999	1051304	16958	7269	233790	1.10	4.497
2000	1140849	9882	7354	267856	1.30	4.3
Veladero						
2005	1883020	53526	4094	61426	0.72	30.65
2006	2573117	60715	13672	560514	2.02	4.59
2007	2732761	51640	17787	518832	0.93	5.27

Table A.7d. Barrick Gold Data (underground)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Hemio						
2002	554146	4006	3458	590131	5.11	0.939
2003	552976	4004	3576	587937	5.14	0.940
2004	530758	4892	3663	541867	4.46	0.979
2005	517581	4496	3504	504573	4.11	1.026
2006	469026	4745	3355	449728	4.05	1.043
2007	509836	5176	3035	370751	3.67	1.375
Plutonic						
2000	1717575	8922	3036	278220	2.91	6.173
2001	1347880	9098	3172	316301	3.12	4.261
2002	1288946	10764	3204	336747	3.33	3.828
2003	1387759	10612	2731	366305	4.22	3.789
2004	1696416	891	2415	334554	4.46	5.071
2005	1362265	10631	1818	275321	4.80	4.948
2007	1344301	996	1852	228155	3.87	5.892
Lawlers						
2003	150459	369	731	108593	4.4	1.386
2004	191095	1325	786	120659	4.5	1.584
2005	174005	2247	806	143694	5.1	1.211
2006	213842	4061	826	120659	4	1.772
2007	243648	513	791	126143	4.7	1.932
Tuluwaka						
2006	646436	7004.88	500.256	153565	9.15	4.210
2007	746200	7341.84	430.272	195874	13.7	3.810

Source:

Mudd, G.M., Global trends in gold mining: Towards quantifying environmental and resource sustainability?, *Energy Policy*, 32, 42-56, 2007.

Annual and Site Responsibility Reports. Barrick, TD Canada, available: http://www.barrick.com, webpage accessed October 20, 2010.

Table A.8.a Anglo Ashanti Data (open pit)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Navachab						
2005	227524	2522	1222	88528	2.1	2.570
2006	299142	6339	1490	94876	1.8	3.153
2007	320274	5786	1597	88034	1.6	1.784
Cripple Creek/Victor JV						
2005	1328301	28482	19194	361628	0.62	3.673
2006	1299069	31969	21795	311010	0.54	4.177
Geita						
2005	1899936	48031	6078	672740	3.14	2.824
2006	3276803	54033	5691	338169	1.68	9.69
2007	3351884	58140	5066	358555	2.01	9.348

Table A.8.b Anglo Ashanti Data (underground)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Sunrise Dam						
2002	2083928	39232	3407	419431	3.5	4.968
2003	2248862	56739	3564	392273	3.1	5.733
2004	2294075	29531	3673	449728	3.5	5.101
2005	2149981	19321	3625	498683	3.7	4.311
2006	2261654	16415	3967	509652	3.6	4.438

Source:

Mudd, G.M., Global trends in gold mining: Towards quantifying environmental and resource sustainability?, *Energy Policy*, 32, 42-56, 2007.

Table A.9. Newmont (open pit)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Inti Raymi						
2005	471118	4046	11532	106838	0.446	0.673
Yanacoocha						
2005	7228549	65580	133036	3656067	0.960	1.977

Source:

Mudd, G.M., Global trends in gold mining: Towards quantifying environmental and resource sustainability?, *Energy Policy*, 32, 42-56, 2007.

Table A.10. Kinross (open pit)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Fort Knox						
2009	2224303	36	16224	263260	0.45	8.449
2008	2217271	1455	13769	329105	0.45	6.737
2007	1,018,567	21700	12722	338459	0.45	5.690

2006	1941970	31751	13462	333383	0.45	5.825
2005	1,049,708	40061	13050	329320	0.45	5.985
Paracuta						
2009	2802322	2290	39744	354396	0.41	7.907
2008	1183322	160	20307	188156	0.41	6.289
Maricunga						
2009	981609	10988	15613	233585	0.57	4.202
2008	852490	10793	15027	221882	0.57	3.842
2005	405,575	6626	5800	67086	0.328	7.921
2006	474,607	12009	14721	256384	0.494	2.924
2007	467,958	11467	13691	246582	0.511	3.027

Source:

Kinross Corporate Responsibility Report, 2009, Toronto, ON, available : www.kinross.com, webpage accessed October 20, 2010.

Table A.11a. Placer Dome (now Barrick Gold) (open pit)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
La Coipa						
2009	978448	10434	4907	231169	1.3	4.187
2008	947397	7889	4918	226293	1.3	2.6
2007	504233	5457	3546	197554	1.3	3.46
2006	537247	877	5126	155180	1.3	4.39
2009	553251	11826	6496	125991	1.3	1.636
Misma						

1997	2080958	24998	5357	235836	1.4	8.824
1998	2044835	25761	5103	203930	1.3	10.0
1999	2132561	32139	6243	217325	1.1	9.813
2000	2156785	23432	6086	238863	1.2	9.029
2001	1688852	2786	5738	182746	1.0	9.2

Table A.11b. Placer Dome (now Barrick Gold) (underground)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Porgera						
1997	3048238	60800	4382	781754	6.9	3.899
1998	3961233	61700	5748	797234	5.3	4.969
1999	4121420	56200	5604	827887	5.4	4.978
2000	4137803	67700	6022	998652	5.8	4.143
2001	4468322	68300	5762	834324	4.9	5.356
2002	3501560	57200	4874	809003	5.2	4.328
2003	4251143	61400	5656	934468	5.3	4.549
Hently						
2001	44640	62.965	196.855	100967	15.1	0.442
2002	60554	133.674	224.252	88919	12	0.681
2003	51480	153.932	289	111960	11	0.460
2004	39319	63	288	156926	16	0.251
2005	38134	91.963	299	122055	12.4	0.312
2006	40616	91.008	308.448	78977	7.9	0.514
2007	48630	102.5	293.9328	76783	8.1	0.633

Source:

Mudd, G.M., Global trends in gold mining: Towards quantifying environmental and resource sustainability?, *Energy Policy*, 32, 42-56, 2007.

Table A.12. GoldCorp (open pit)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Wharf						
2004	341260	9234	3036	83495	0.87	4.087
2005	290884	11357	3006	68524	1.03	4.245

Source:

Warf Resources, INC. 2005 Sustainability Report, Lead, SD, 2005.

Table A.13. GoldFields (open pit)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Damang						
2004	880,300	9855	5236	338204	1.8	2.603
2005	870,800	9050	5215	271685	1.5	3.205
2006	1,048,522	21427	5328	257894	1.4	4.066
2007	1,538,382	28110	5269	206083	1.1	7.465
2008	1,325,361	29433	4516	213066	1.3	6.220

Source:

Annual Report, Gold Fields Limited, Years 2008 and 2009, Johannesburg, Gauteng, available: www.goldfields.co.za, webpage accessed October 20, 2010.

Table A.14. Rio Tinto (underground)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Kelian						
2002	3204523	18087	7313	591227	3.0	5.420
2003	2496795	2018	7853	514445	2.4	4.853

Source:

Mudd, G.M., Global trends in gold mining: Towards quantifying environmental and resource sustainability?, *Energy Policy*, 32, 42-56, 2007.

Table A.15. Teck (underground)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (oz)	Grade (ppm)	GJ/oz prod
Pogo						
2006	514084	187	287	124349	14.5	4.134
2007	616520	190	649	284996	15	2.163

Source:

Mudd, G.M., Global trends in gold mining: Towards quantifying environmental and resource sustainability?, *Energy Policy*, 32, 42-56, 2007.

Complete Listing of Copper Mine and Mill Data

Table A.16. BHP Billiton (open pit/co-producing)

Year	GJ(e+t)	Waste(kt)	Ore Milled (kt)	Produced (t)	Grade (ppm)	GJ/t produced
Escondida						
2003	12543060	230035	84049	993,000	13700	12.63
2004	13816085	294995	98021	1,207,600	14300	11.441
2005	13910700	273515	102984	1,270,000	14200	10.953
2006	14412709	254425	140222	1,313,000	10900	10.977

Source:

Mudd, G.M., Personal Communication. October, 2010.

Table A.17. Teck (open pit/co-producing)

Year	GJ(e+t)	Waste (kt)	Ore Milled (kt)	Produced (t)	Grade (ppm)	GJ/t produced
Highland Valley						
2007	4972982	27901	42593	139500	3700	35.6486191
2006	4772990	14905	45356	414230	4100	11.522561
2005	4638969	12070	50666	459051	4000	10.1055640
2004	4465965	15214	50623	453214	3800	9.853988

Source:

2007 Sustainability Summary, Highland Valley Copper Operations-Teck Cominco, Vancouver, Canada, available: www.teck.com, webpage accessed October 20, 2010.

Table A.18. Lundin Mining (open pit/co-producing)

Year	GJ(e+t)	Waste (kt)	Ore Milled (kt)	Produced (t)	Grade (ppm)	GJ/t produced
Aguablanca						
2009	135000	3520	561	6989	4000	19.316
2008	117249	4435	396	7071	4000	16.582
2007	37991	1220	143	6281	5000	6.049
Neves Corvo						
2009	822273	2332	2570	86462	39000	9.510
2008	786388	2344	2410	89026	43000	8.833
2007	663199	1620	2181	90182	48000	7.354

Source:
Lundin Mining 2009 Sustainability Report, Toronto, ON, available: *www.lundinmining.com* webpage accessed October 20, 2010.

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Vita

Emily Loree Tavrides was born and raised in Lakeland, Florida and attended college at the College of Charleston in Charleston, SC. After receiving her Bachelors of Science in Physics and mathematics, she worked in biomedical engineering at the Medical University of South Carolina, interned with a medical physicist, and worked alongside an oceanographer in Tampa, Florida. After exploring many research options she committed to furthering her education at the University of Texas at Austin in Fall 2009. She has since been in pursuit of her master's degree in UT's Nuclear and Radiation Engineering Program. Emily is a world traveler and outdoor enthusiast, currently residing in Austin, Texas.

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